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MINING PRACTICES, METHODS, AND COSTS AT ELKORO MINES, JARBIDGE, NEV.

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DEPARTMENT OF COMMERCE - BUREAU OF MINES

MINING PRACTICES, METHODS, AND COSTS AT ELKORO MINES, JARBIDGE, NEV.[1]

By John Furness Park[2]

INTRODUCTION

This paper, which describes the mining methods and practices at the Jarbidge mines of the Elkoro Mines Co., is one of a series of papers being issued by the United States Bureau of Mines on mining methods and costs in various districts throughout the United States.

LOCATION AND HISTORY

The Elkoro Mines Co.'s property is located at Jarbidge in the northeast corner of the State of Nevada (fig. 1). The camp is 9 miles south of the Idaho line; the nearest railroad point is 70 miles to the northeast at Rogerson, Idaho. A road leading to the south and connecting with Deeth, Nev., is passable during the summer months; the heavy snowfall which generally takes place early in the winter usually closes the road until the first of July.

The town of Jarbidge is situated in a narrow canyon at an elevation of 6,100 feet above sea level. The walls of the canyon rise steeply on each side of the camp, protecting it somewhat from very severe weather and making possible a system of mining by adits.

The original discoverer of the camp, D. A. Bourne, entered the region from the west on a prospecting and fishing trip in 1909. In panning the gravel along the creek bottom he found free gold which apparently had its source on the hillside to the east of the main valley. He staked a number of claims before returning to civilization, and by his reports created a small gold rush. Within a few months a tent town of about 2000 persons had been built in the canyon, and the hillsides for several miles around were located. The next 4 or 5 years proved to be a period of extensive speculation but rather meager development, and the camp gradually dwindled to a population of 400 or 500. Three or four mills had been built during the boom times, but the orebodies were mostly too small to warrant any such investment, and the stockholders in these ventures lost a great deal of money.

During 1915 and 1916 a number of claims were consolidated under one company and a vein was opened up which proved to be of commercial value. This property was acquired by the Elkoro Mines Co., by whom a mill was constructed in 1917. This company has operated continuously since that time, with an average mine production of between 100 and 150 tons of gold-silver ore per day.

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1 The Bureau of Mines will welcome reprinting of this article provided the following footnote acknowledgment is used: "Reprinted from U. S. Bureau of Mines Information Circular 6543."

2 One of the consulting engineers, U. S. Bureau of Mines, and mine superintendent, Elkoro Mines Co.

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GEOLOGY AND PHYSICAL CHARACTERISTICS

The Jarbidge mining district consists of a series of Tertiary volcanic flows which have undergone extensive alteration. They are cut by a large number of faults of varying degrees of magnitude and movement which make the orebodies exceedingly difficult to follow. The gold and silver is found in quartz veins having average dips of 60 to 80° from the horizontal. The strike is roughly southeast and northwest but is subject to local changes edue to extensive ground movement.

The veins vary greatly in length and width. The principal vein which has been developed in the district has been followed in ore for a distance of over a mile and no doubt extends in some form for several times that distance. Other veins, principally of a grade too low for mining, can be traced on the surface as far as 4 or 5 miles. The width of the main vein averages about 6 feet, ranging from narrow stringers up to 20 feet in rare instances.

The vin is enriched by small leads usually entering and crossing the main ledge at about a 60° angle. At these places it is often possible to widen the stopes slightly. Experience has shown that these branching or intersecting veins do not carry commercial values away from the main ore body.

An operating difficulty is usually experienced in mining the ore of split-off shoots extending into the hanging wall. The values may extend 3 or 4 feet along the diverging vein. In stoping out this ore, a shelf is formed in the hanging wall that requires a large amount of timbering and blocking.

Except at the crossings the walls of the veins are clearly defined. At these points some of the wall rock usually is broken to make sure that all the commercially valuable material is obtained. Fault zones roughly paralleling the vein and located 6 to 8 feet in the hanging wall occur at a few places in the mine.

> The economic metals of the deposits are gold and silver. They occur in the form of native gold, gold-silver alloy, or electrum, and the silver-bearing minerals argentite, cerargyrite, and naumannite. The primary metallization produced mainly sulphides, chiefly argentite and pyrite, but from the oxidized portions of the veins pyrite has practically disappeared.
>
> The chief gangue minerals are quartz and adularia, and a common feature is their microcrystalline to cryptocrystalline character. The quartz is the more important, as its dominance or abundance favors the occurrence of the ore minerals, especially gold.[3]

No ore of very high value is found, a period of years yielding a mill feed of slightly over $10. The values occur fairly regularly through the vein; that is, there are no pockets of ore, although here and there along the vein lean spots occur which, in stoping, are usually left as pillars.

During the early years of mining at Jarbidge the amounts of gold and silver recovered were about equal, by weight. As operation continued into veins further to the west, however, the proportion of silver has increased, until at present there is about ten times as much silver as gold in the finished bullion. These workings are still in the oxidized zone.

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3 Schrader, F. C., The Jarbidge Mining District, Nevada: U. S. Geol. Survey Bull. 741, 1923, p. 26.

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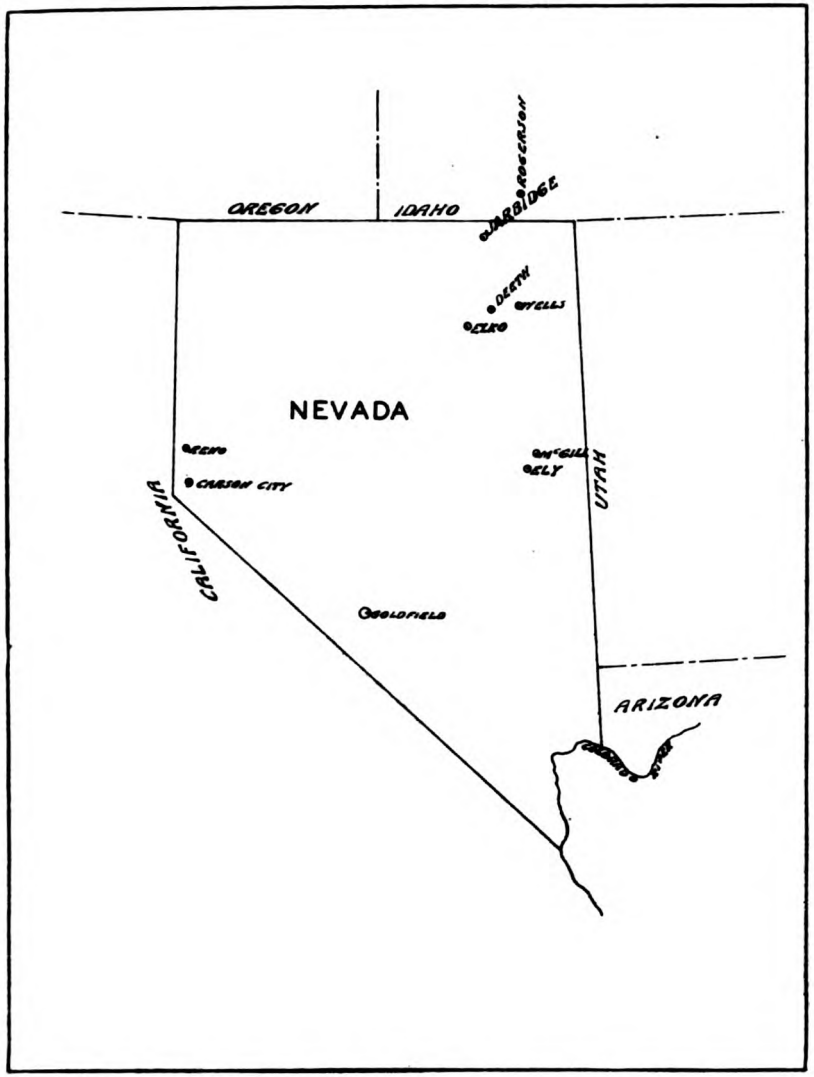

Figure 1.- Location of Jarbidge and other mining camps

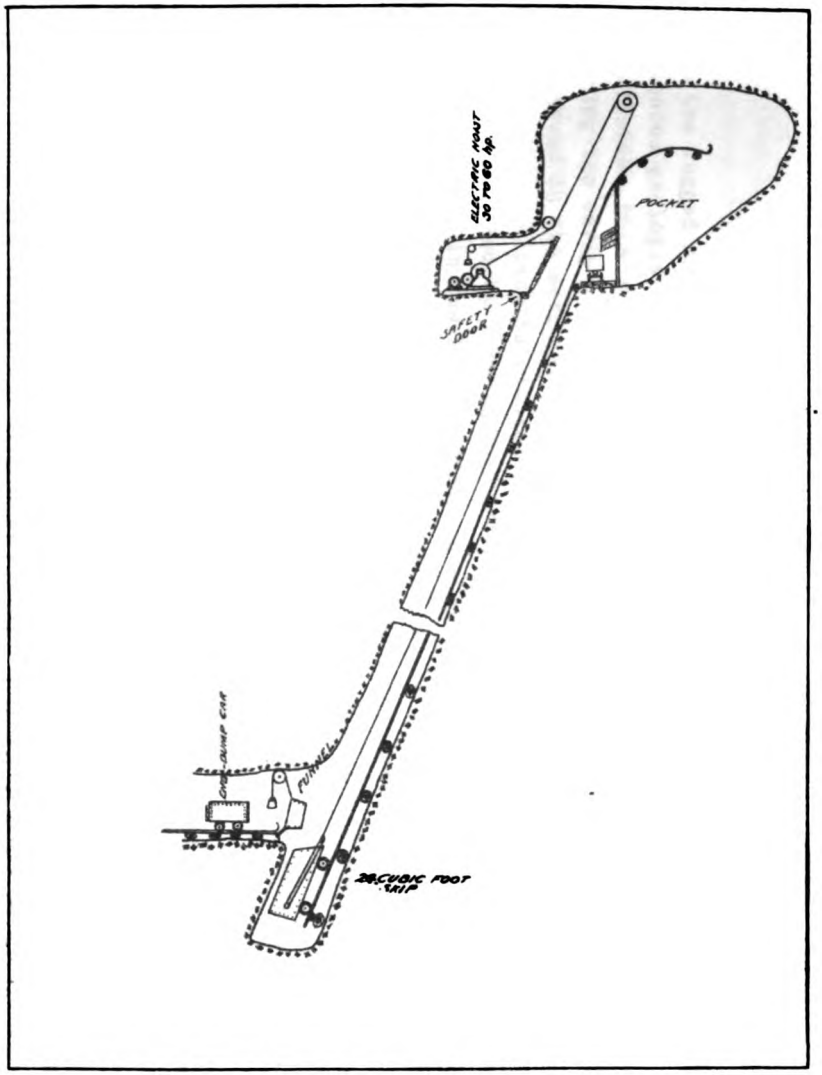

Figure 3.- General arrangement of winze

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PROSPECTING AND EXPLORATION

Although the rugged nature of the country exposes numerous outcrops which are of value to the experienced geologist, the overburden, consisting principally of gravel and slides, covers the hillsides to a sufficient depth to hide the more obvious indications of vein material. During the early years of the camp, attempts were made to locate veins by a study of the values found in the surface covering. Test holes were dug at regular intervals, the dirt was panned, and the results plotted with the purpose of locating any concentration of values. The data obtained by this method proved to be unreliable, for, although gold and silver values were found in the loose overburden in the vicinity of known veins, a number of locations which showed even better pannings proved not to be associated with ledges of any kind. This condition was no doubt due to the steepness of the country as a whole, which would cause movement of the loose material for long distances without any definite trail to indicate the original source.

The Elkoro Mines Co. has taken the obvious course of having the surface geology carefully mapped in detail and a study made of all openings available over a fairly wide area. This work was done by a geologist who was thoroughly familiar with the type of formations encountered and whose conclusions have proved extremely valuable in the constant search for new orebodies.

A careful study of the rhyolite flows led to the conclusion that the veins are of value only within certain vertical limits outside of which the gold and silver content falls below the point of commercial value, although the vein itself may continue and be as strong and well defined as before. This vertical component seldom exceeds 650 feet, so that the problem of finding ore beyond any large fault zone involves not only relocating the vein itself but finding it in a flow which is favorable for the ore. This is a serious difficulty because the movement on the fault planes often exceeds a thousand feet vertically, horizontally, or both, and the rhyolite flows are very similar in appearance and characteristics.

The combination of these features tends to make prospecting and development both slow and expensive. The presence of faulting throughout the orebody, moreover, causes weak walls and spots with low mineral values, thus having a detrimental effect on ore extraction.

Diamond drilling has been done, but with only slight success. Not only are the gold and silver values irregularly distributed in the veins, but the vein material cores poorly. As a result, the numerous fault zones penetrated by the drill produce sludge which is difficult to distinguish from true vein material, and the difficulty is increased by the minor gold and silver values frequently found in such zones. Diamond drilling is consequently unsuitable as a prospecting medium. However, it might prove very useful as a means of determining the thickness, character, and relations of the various rhyolite flows in new areas.

SAMPLING AND ESTIMATING TONNAGES

Sampling is done by hand, following the ordinary practice of cutting a shallow channel across the face or back.

Development headings are sampled after each round. In stopes, cuts generally consisting of from 3 to 5 samples are made across the entire width of the stope at intervals of 5 feet throughout the length. By careful inspection of the assay results of the samples thus obtained it is possible to control the width of the opening to the best advantage.

In estimating orebodies the results of the drift samples are averaged, together with those from any raises or winzes which may pass through the ore. The widths and values thus obtained are assumed to be correct for the block. The cubical contents of the block are calculated and reduced to tons by the use of a factor which varies from 17 to 19 cubic feet per

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ton of ore in place. This figure is one which has been found by experience to make the proper correction for variations in stoping widths.

The value per ton estimated for any block of ore by this method will average about 17 per cent higher than the actual extraction, but the difference is more than compensated for by the increase in actual tonnage over that estimated. Ordinarily the total number of tons obtained from an orebody will exceed the estimate by about 20 per cent, due partly to the additional material broken as explained before, and partly to dilution caused by the sloughing of the walls during the drawing of the stopes. These factors are considered when reporting estimates, but it is almost impossible to estimate any one block of ore within 20 per cent of its final production. This is due to the continually changing width of the vein and the changes in the nature of the wall rock, which cause more or less dilution of the broken ore.

DEVELOPMENT AND MINING

No definite plan of development can be decided upon in advance, owing to the irregularity of the orebodies and the necessity of adapting the operations to local conditions. Up to the present it has been possible to open all the workings with adits from the surface; the ore is transported to the mill either by train or by aerial tramways. However, the company has recently opened up a body of ore below the level of the lowest adit and the material from this level is hoisted through a winze to a haulage aidit

The adits from the surface are spaced at approximately 150-foot vertical intervals and two of them serve as main haulage ways. The ore is either hoisted to one of these levels or run through ore passes from higher stopes (fig. 2).

All the levels are connected by raises or winzes at convenient points. These are usually driven to explore orebodies but are so placed as to serve later for stope manways or ore chutes. They also provide ventilation for stoping.

Winzes

Winzes are of the 2-compartment type and are sunk either in the vein or in the wall rock, depending on which will be more convenient for the work in view. In either case they are on a dip of approximately 70° from the horizontal which has the advantage of causing slight wear on the wheels and running gear of the skips owing to the comparatively light pressure against the walls. Each compartment is 4 by 4 feet in the clear, one side accommodating the skipway and the other the ladders, air pipe, and other equipment.

At the upper end a skip pocket is construoted which holds from 75 to 100 tons (fig. 3). It has been found advisable to make such pockets large enough so that when hoisting is done on three shifts the pockets need be drawn only on the day shift. On the lower level the ore is trammed by hand and dumped directly into the skip in the winze. Because of the short hoisting distance the skip can be pulled up, dumped, and returned in such a short time that no delay is experienced by the trammers.

To avoid excessive spilling around the sides of the skip when dumping into it, a square funnel is hinged to the edge of the track level in such a way that the car in approaching the skip knocks it into position for dumping. The trammer after dumping his car pulls a rope which raises the funnel to the upright position again where it clears the skip.

As the winzes seldom exceed 150 feet in depth the sinking buckets and ore skips are of small capacity. Ordinarily a round bucket of 1,200 pounds capacity is used during the sinking operations; this runs on a skidway of round poles of from 6 to 8-inch diameter. When the winze is to be used to hoist ore from stopes, a track is substituted for the pole skids

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and the bucket is replaced by a self-dumping skip with a capacity of 24 cubic feet. It has been found necessary to build the box of the skip from extra heavy material to withstand the continual dumping from cars. This has proved to be much more damaging than loading from a chute, as there is a tendency for trammers to place large rocks on top of their loads, which naturally strike the bottom or back of the skip before there is any protecting covering of fines.

The winze hoists are of 30 to 60 horsepower, and use ¾-inch wire rope. They are placed in the hanging wall at the level of the winze collar, thus allowing the hoist operator ready access to the skip when loading supplies.

Drifts and Crosscuts

Drifts and crosscuts are driven 5 by 7 feet in the clear and are timbered with round native pine timber. Sets are placed at 5-foot intervals and lagged with either split poles or 3-inch plank cut at a local sawmill. During the past few years a shortage of available pine timber for saw logs has caused the company to resort to the use of a cottonwood found along the canyon. This wood has proved to be extremely tough and serviceable in the mine, although when green its weight makes it somewhat troublesome to handle.

The ground will generally stand for some time without timbering except in faulted sections, but in drifts or tunnels which are used for extended periods the ground tends to air-slack to such an extent that it is usually necessary to place sets and lagging throughout the greater part of their length.

Drifting is generally done on two shifts, the crew consisting of one miner and two muckers for each round. The muck is taken out by hand during the drilling of the round. It is seldom necessary to timber close to the face during the driving so that the timber crews are able to work well back of the working face and independently of the drilling crew.

No attempt is made to use standard rounds, as the condition of the rock varies greatly from one round to the next. Ordinarily from twelve to sixteen 6-foot holes are drilled, using a down cut. One-man drifting machines are used which take 1⅛-inch hollow round steel. The bits are formed with 1/8 inch changes in gage. All blasting in the mine is done with 40 per cent strength, low-freezing gelatin dynamite in 1¼ by 8 inch sticks and No. 8 caps. No stemming is used. The miners blast their own rounds and report to the shift boss any missed holes so that the next shift will be properly warned before going to work.

Raises

Raises are generally run in the vein and the full width of the ore, which seldom exceeds 12 to 14 feet. The length of the raise along the vein will be from 10 to 20 feet, depending on what use the raise will be put to in the future. In cases where it will serve as a stope manway the length is increased so that a chute can be built on each side of the manway. On the other hand, if the raise is put up in a low-grade section of a vein in hope of finding milling ore above, it is made as small as working conditions will allow.

Raises are timbered with round stulls spaced 5 feet apart and lagged. In order to facilitate the handling of timber and drilling supplies in the raises a narrow slide of 2-inch plank in which a small skip is operated is carried up alongside the ladder. The hoisting is done by means of small air hoists of the "tugger" type, which are easily and quickly installed at the foot of the manways.

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STOPING

Two methods of stoping have been used at Jarbidge, the shrinkage and the cut-and-fill methods. The greatest part of the ore taken out has been mined by the first method and the filling of stopes has been resorted to only when the wall rock was extremely unstable. During 1930, to which year most of the costs given in this paper apply, no filled stopes were worked. It is probable, however, that at least one-third of the ore produced in 1931 will be taken from this type of stope.

Shrinkage Stoping

In starting shrinkage stopes, drifts the full width of the ore are run on the vein and a round is shot out of the back. Stulls are placed and the drift lagged over through the section to be stoped. Chutes for drawing the ore are spaced at 15-foot intervals. These are built of 3-inch native lumber and are not standardized, as it has been found that better results are obtained when the chutes are cut and fitted where they are to be used. In the building of chutes the cottonwood lumber mentioned previously has proved very serviceable because of its long-wearing qualities. Manways are placed at intervals determined by the conditions encountered or by raises which have been put up for development purposes and can be utilized for the stopes. The manways will generally average 125 feet apart, the determining factor being the distance which the hose for the stoping machines must reach to the nearest air line. In order to simplify the connecting of a number of machines at some distance from a manway, a large hose is connected to the air pipe and run across the stope where it ends in a manifold to which the individual machines may be attached. These large hose are 50 feet long and are much quicker and easier to connect than a temporary pipe.

The length of a stoping section is determined by the distance between two manways, and is, therefore, usually about 125 feet. Each section is usually carried up horizontally, but adjacent sections are not necessarily kept abreast, and their relative advance is determined by operating convenience.

Drilling is done with stopers. One-inch hollow square drill steel with a double-taper crossbit is used with all wet stopers. The vein material is drilled rather easily, as it is of only medium hardness, and it is seldom that a driller will use more than three sets of steel during a day. The work is done on the basis of a day's pay in the stopes, since the size and shape of the vein is constantly changing and it is difficult to make any definite standard for contract drilling. A bonus system based on the number of feet drilled was put in operation at one time, but it was found to be unsatisfactory, as the holes were not as carefully placed and the resulting loss in explosives more than offset any gain in drilling.

Very little plugging is done in the stopes. Such plugging as is necessary at the chutes is done by the trammers, who use stoping drills from the floor of the drift.

During the breaking of the ore about 23 per cent is drawn from the stope, which leaves the top of the broken ore at the proper working distance from the back. Very little difficulty is encountered in the form of ore arching or otherwise hanging up in the stopes while drawing. Usually when a chute fails to run, drawing an adjacent chute will bring the ore down without damage. More obstinate chutes are brought down by exploding a small amount of powder on the end of a long stick.

Shrinkage stopes are run up to as high as 250 feet, with very little timber. Pillars are left at lean points in the vein or where the wall shows signs of weakness. When the breaking has been completed (usually about 4 feet below the next level), a strong overhead bulkhead of poles is placed through the entire length of the stope to protect the timber crews who follow down the ore as it is drawn. These men place stulls across the stope to

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support all loose or weak portions of the hanging wall and thoroughly clean down the walls as the ore recedes. Any pillars of valuable ore are removed if it is possible to replace them with timber. Although this method does not leave the stopes in a permanently safe condition, as the timbers rot in a matter of a few years' time, the subsequent caving is gradual and not particularly harmful to working levels.

The high timbering-labor charges shown in the cost tables are due chiefly to cost-keeping methods. All labor in stopes, except actual drilling and blasting, is done by timbermen and their helpers. Much of this, especially during the drawing off of stopes, is not timbering but mucking, yet because of the difficulty of segregating each man's time to several operations, it is all charged as timbering.

Cut-and-Fill Stopes

The filling of stopes, as explained before, is done only where conditions make it absolutely necessary. The principal occasion for filling is the occurrence of fault zones in the hanging wall, as previously mentioned, which causes too great a weight on the open stope to be supported by timber. As these conditions are well understood, filled stopes instead of shrinkage stopes are always started in such places, and in no case has it been necessary to convert a shrinkage stope into one of the cut-and-fill type.

Filled stopes are provided with cribbed ore chutes, the cribbing of sawed lumber being framed on the surface. The chutes are close enough together so that the ore can be shoveled directly into them by hand, and no cars or scrapers are used in the stopes.

Waste fill is derived from waste raises driven for this purpose at convenient points in the walls of the stope, and from development work on upper levels. The latter is dropped through the ventilation raises directly into the stope and spread by hand. As these raises are usually only one or two in number in any section, this source of fill serves only a minor part of the stope. Deflecting chutes are sometimes built under the raises to increase the spread of the ore.

In filled stopes the ore is blasted onto shoveling floors of 2-inch fir plank, which prevent excessive loss of fine ore in the waste fill.

UNDERGROUND AND SURFACE TRANSPORTATION

Track of 12-pound rails and 18-inch gage is used throughout the mine. The ore is drawn by trammers from the stopes into 20-cubic foot end-dump cars. These are trammed either by hand or in short trains by mule power and dumped into the skips, on lower levels, or through grizzlies with 12 by 12 inch openings over the ore passes on upper levels.

On the present main haulage level ore is drawn from stopes, ore passes, or winze pockets into 26-cubic foot rocker-dump cars. Trains of 8 cars are hauled by a 2½-ton storage-battery locomotive over a distance of 1,600 feet to the mill bin.

At a level 350 feet above this main haulage level the ore is trammed to the surface in cars of a similar type and dumped into a bin. From this bin it is transported by aerial tramway to the mill.

The tramway is 1,200 feet long and uses eight 645-pound capacity buckets. The speed ranges between 275 and 300 feet per minute, being controlled by a 15-hp. electric motor, which serves chiefly as a brake. The operating cost over a period of years has been 13.54 cents per ton. During periods when the company was operating other small mines in the vicinity, as many as four aerial tramways served as feeders for the main tramway. Two of these smaller lines were of the 2-bucket type, using 1,000-pound buckets, and had capacities of about 100 tons per 8-hour shift.

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The lower principal adit is at the elevation of the townsite, so that most of the timber and supplies for the mine are taken in through this adit. An inclined tram line on the surface with a 30-hp. electric hoist is used to transport such material as is necessary to the higher levels.

COMPRESSED AIR, VENTILATION, DRAINAGE, AND SAFETY

One compressor is located at the townsite adit level and another at the principal working level for the higher workings. The compressores are connected so that either one may serve the whole mine or they may be run together. This is a decided advantage, as for short periods the air load exceeds the capacity of one machine while usually either compressor will supply all the air required. The blacksmith shop is at the adit level.

Natural ventilation is sufficient for the whole mine, since all levels are connected by raises or winzes. In development tunnels the smoke from blasting is removed by the use of a small amount of compressed air after the round is shot. In the case of long drifts or crosscuts where considerable dead air may accumulate during the driving, an electrically driven blower of the propeller type is placed at the nearest point where there is a circulation of fresh air, and a 12-inch air duct of flexible tubing is hung along the back of the tunnel level to carry the air to the face. One or two of these blowers are always kept on hand at the mine, with a supply of tubing, so that they are ready for installation at short notice.

During the last year a block of ore has been developed which lies below the level of the bottom of the canyon. It was obvious that considerable water would be encountered, and the following method has been worked out and put into operation. A winze was sunk in a particularly solid rhyolite formation in the footwall of the vein. The rhyolite not only provided strong walls in the winze itself but also tended to hold back the water during the sinking operations, so that the work was completed with the use of a single electric sinking pump in conjunction with an air sinker of fairly small capacity.

From the bottom of the winze a crosscut was run for 70 feet in a direction opposite to that which would be taken by the working crosscut connecting with the vein (fig. 4). From this point a vertical raise was put up to the level of the collar of the winze, where it connected with a crosscut which had already been driven on this upper level. A deep-well pump of a 1,000-gallon per minute capacity was installed in the vertical raise and was thus independent of the main hoisting winze. The crosscut to the ore was started at an elevation above the back of the previously mentioned water crosscut so that there was at all times a reserve water capacity equal to the water crosscut. This would probably permit a shutdown of two or three hours, which should allow time for minor repairs to the pumping equipment or the adjustment of a power failure if not too serious. An additional advantage is the fact that by the use of baffles the water in passing through the 70 feet of the crosscut can be cleared of sharp particles of grit which would otherwise greatly increase the wear on the pump runners.

The installation of this pump has been complete for only a few months so that no data are as yet available regarding its running or upkeep costs.

The mine receives its electric power over a high-tension power line from a hydroelectric plant in Idaho approximately 80 miles distant. As this line passes through an uninhabited section of sagebrush country there is always the possibility of shutdowns extending to 50 or 60 hours. This was the deciding factor in the choice of the type of pump to be used. The deep-well pump gives every indication of being entirely satisfactory, since in case of a

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shutdown there is no delay in starting up when the power is restored and no harm is done to the pump itself.

These pumps operate quietly and with practically no vibration; with proper care they should give as good service as other types of centrifugal pumps.

The pump installation is so designed as to permit the addition of a second pump of equal capacity in the raise, if necessary.

No organized fire-fighting or rescue crews are maintained, as the risk of fire in the mine is small, owing to the natural dampness of the workings. The mine is kept clean and no inflammable material is allowed to collect.

A fair proportion of the men employed have received the course of first-aid treatment given by the United States Bureau of Mines, and the proximity of the mine to the company hospital permits prompt medical assistance in cases of injury.

TRUCKING SUPPLIES FROM ROGERSON TO JARBIDGE

All supplies are brought into Jarbidge by motor truck from Rogerson, Idaho. This 70-mile haul is over a dirt roadway, partly through uninhabited sagebrush country and partly following the winding canyon of the Jarbidge River. The road is occasionally closed for a few days during winter months, but no other serious difficulty has been encountered in the past three years. Because of the many sharp curves and some steep grades, trucks of rather small capacity are used.

This freighting is contracted for at a price now slightly less than 1 cent per pound. Five years ago the price was fixed at 2 cents, but by a gradual improvement of the road and the elimination of the worst grades at the expense of the mining company, the present rate has been made possible.

COSTS

Tables 1, 2 and 3 of costs and unit production cover in general the year 1930, with a few exceptions where figures are taken for longer periods. The tonnage shown is that trammed but the amount broken in the mine was within a few tons of the same amount.

Table 1.- Operating costs, exc luding development

Mining method, Shrinkage. Tonnage trammed, 57,539.

	Labor	Super-vision	Compressed-air, drills, steel	Power	Explo-sives	Timber	Other supplies	Total
Mining	$0.382	-	$0.048	[1]$0.114	$0.107	-	$0.208	$0.859
Timbering	.537	-	-	-	-	$0.164	.050	.751
Haulage	.295	-	-	.010	-	-	.027	.332
Hoisting	.073	-	-	.008	-	-	.003	.084
General underground	.148	$0.152	-	.007	-	-	.045	.367
Surface	.175	-	-	-	-	-	.057	.217
Total	1.610	0.152	0.048	0.139	0.107	0.164	0.390	2.610

1 Compressed air.

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Table 2.- Details of mining costs on per-ton basis

Mining method, Shrinkage. Tons of ore mined, 57,539.

	Cost per ton	
Breaking (drilling and blasting)		$0.872
Labor	$0.382	
Supplies (not including explosives)	.208	
Explosives	.107	
Warehouse handling charge	.006	
Compressed air	.114	
Surface transportation	.007	
Steel and tool sharpening	.047	
Machine repairs	.001	
Timbering		.796
Labor	.537	
Supplies	.050	
Timber and lumber	.164	
Warehouse handling charge	.004	
Surface transportation	.012	
Tool sharpening	.029	
Tramming		.356
Labor	.295	
Supplies	.027	
Surface transportation	.007	
Blacksmith-shop charges	.017	
Power	.010	
Hoisting		.088
Labor	.073	
Supplies	.003	
Power	.008	
Repair charges	.004	
Sampling		.087
Labor	.024	
Assaying	.060	
Surface transportation	.003	
Ore sorting		.014
Miscellaneous labor charges		.009
Miscellaneous supplies		.005
Superintendency		.152
General		.101
Engineering		.069
Power (miscellaneous)		.007
Hospital expense		.034
Housing and feeding expense		.020
Total charges to mining		2.610

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Table 3.- Operation costs in units of labor, power, and supplies[1]

Labor (man-hours per ton):	0.555
Breaking (drilling and blasting)	0.555
Timbering	.818
Haulage and hoisting	.577
Supervision	.122
Miscellaneous	.013
Sampling	.035
Total direct underground labor	2.120
General labor chargeable to underground operation:	
Ore sorting	0.023
Engineering	.053
General	.041
Average tons per man-shift	3.576
Labor, percentage of total cost	61.82
Power and supplies:	
Explosives (pounds per ton) 40 per cent gelatin	1.239
Timber (linear feet per ton)	.852
Lumber (board feet per ton)	1.355
Total power (in kw. h. per ton)	10.919
Air compression	8.477
Hoisting	1.845
Pumping	.507
Lighting	.090
Supplies and power, percentage of total cost	38.18

1 Exclusive of development.

Tables 4 and 5 of development costs show the costs per ton and per foot of the three principal types of development openings. The costs per ton are based on the tons of rock or ore actually broken and removed in the course of the development work and bear no relation to the total tonnage of ore removed to the mill. Naturally in the opening up of ore bodies The ore removed returns a profit which is credited to the development work; however, this is not taken into account here.

Drifting and crosscutting are combined as below since the two are similar in sizes and costs, and no variation is shown between development work in ore and in the country rock since practically no difference exists in the costs.

	crosscutting	Raising	Sinking
Size of excavations, feet	6 by 8	6 by 20	7 by 11
Timbered, or not	About 10 per cent	Yes	Yes
Physical properties of rock:			
(a) Hard, or soft	Fairly hard	Fairly hard	Fairly hard
(b) Firm, or loose	Firm	Firm	Firm

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Table 4.- Development costs

	Drifting and crosscutting	Raising	Sinking[1]	Total development
Footage advance	5,266	616	646	6,528
Costs per foot:				
Labor	$ 6.284	$ 5.172	$15.146	$ 7.056
Supervision	.293	.293	.293	.293
Compressed air	.580	.484	.936	.605
Steel sharpening	.577	.667	.315	.531
Power	.171	.090	.434	.217
Explosives	2.094	1.071	2.106	1.999
Timber	.018	1.055	2.762	.388
Other supplies	.286	.450	.799	.353
Total underground	10.303	9.273	22.791	11.443
Outside expense	.351	.351	.351	.351
Assaying	.112	.112	.112	.112
General expense	.565	.565	.565	.565
Total cost per foot	$11.331	$10.301	$23.819	$12.471
Tons removed in development	18,431	4,620	2,665	25,716
Cost per ton	$ 3.238	$ 1.375	$ 5.773	$ 3.166

1 The figures given cover the sinking of development winzes and not permanent shafts, which naturally would show a much higher cost.

Table 5.- Development costs in units of labor, power, and supplies

	Drifting and crosscutting	Raising	Sinking[1]	Total development
Labor (man-hours per foot):				
Breaking (drilling and blasting)	2.741	2.182	5.350	2.946
Timbering	.327	3.325	3.443	.918
Shoveling	5.627	-	5.696	5.103
Haulage and hoisting	.178	2.312	7.443	1.098
Supervision	.319	.415	.310	.327
Total labor	9.192	8.234	22.242	10.392
Feet per 8-hour shift	.870	.972	.360	.770
Labor, percentage of total cost	69.08	63.34	69.71	68.77
Power and supplies (per foot):				
Explosives (pounds)	8.963	6.274	9.177	8.731
Timber (linear feet)	.440	6.002	3.065	1.224
Lumber (board feet)	-	10.063	19.576	2.931
Supplies and power, percentage of total cost	30.92	36.66	30.29	31.23

1 The figures given cover the sinking of development winzes and not permanent shafts, which naturally would show a much higher cost.

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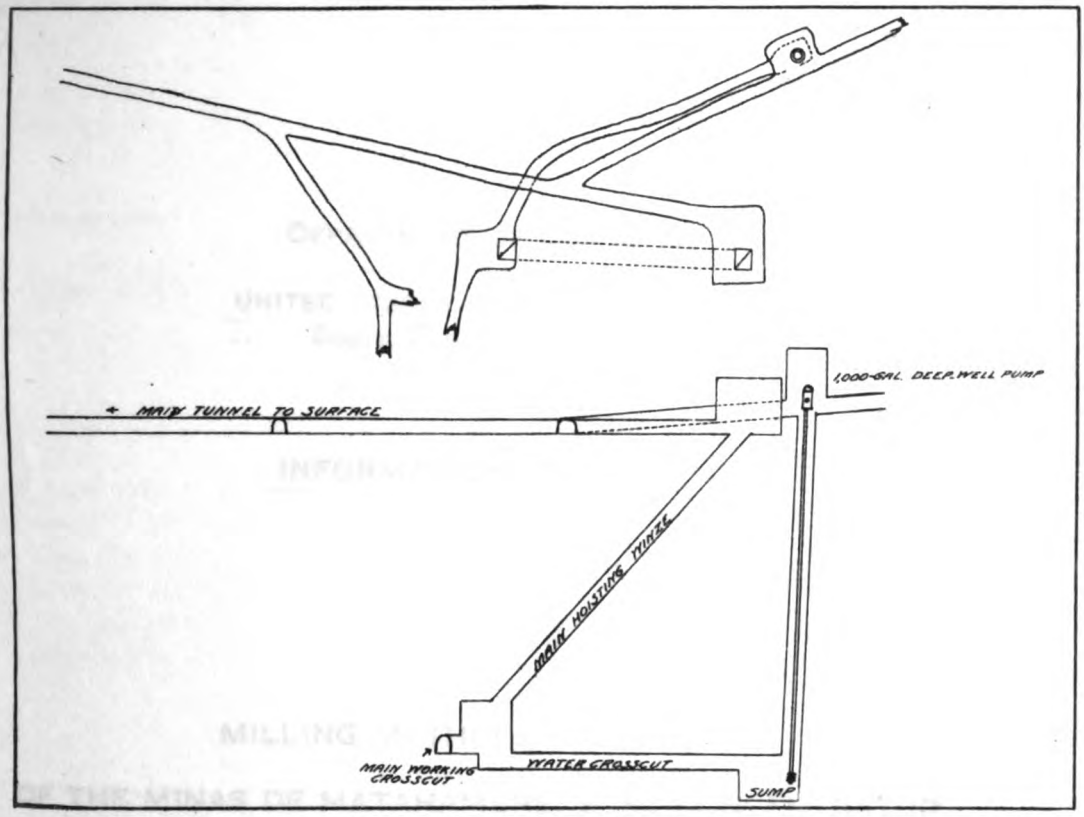

Figure 4.- Plan and elevation of pumping arrangement

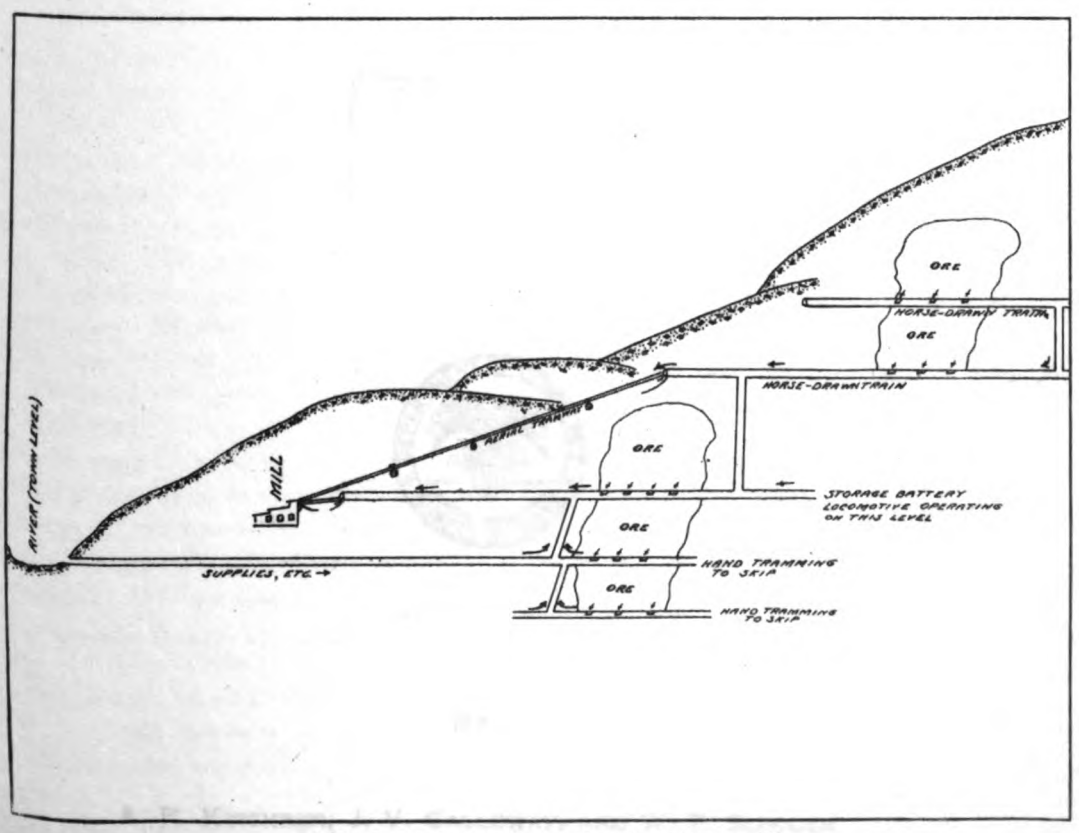

Figure 2.- Projection of direction of ore flow to mill

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